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BUREAU OF MINES
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INFORMATION CIRCULAR

MILLING AND SMELTING OPERATIONS OF THE MAGMA COPPER CO., SUPERIOR, ARIZ.

BY

EDWARD J. CALDWELL

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By Edward J. Caldwell[2]

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2/ Assistant metallurgist, Magma Copper Co., one of the consulting engineers, Bureau of Mines.

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INTRODUCTION

In 1930 the Bureau of Mines published a description3/ of the concentrating process of the Magma Copper Co. as it had been developed to that time and was then employed. The present paper revises the description to the end of 1943 and includes a description of the smelting processes.

Data from the mill and smelter operations for 1942 are given in tables 1 to 14. Milling costs are included, but it is not permissible to publish smelter costs.

ACKNOWLEDGMENTS

Appreciation is expressed to E. G. Dentzer, general manager of the Magma Copper Co., and to J. H. Rose, smelter superintendent, for permission to prepare this paper. The author is indebted to Mr. Rose, to E. E. Slack, plant engineer, and to W. S. Nesbit, gas engineer, for their assistance in obtaining data, and to Dr. M. N. Short of the University of Arizona for the description of ore minerals.

LOCATION

The mine, mill, and smelter are at Superior, Pinal County, Ariz., in the south-central part of the State, approximately 70 miles southeast of Phoenix and 21 miles southwest of Miami.

3/ Rose, J. H., and McNabb, J. C., Milling Methods and Costs on the Concentrator of the Magma Copper Co., Superior, Ariz.: Bureau of Mines Inf. Circ. 6319, 1930, 20 pp.

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The town is served by the Magma Arizona Railroad, a company-operated road with 30 miles of standard-gage track, which connects at Magma with a branch of the Southern Pacific Railroad.

The plant is on the north side of the town. The mine is on the east side of a ridge; the mill is on the west side of the ridge and one-half mile west of the mine. The smelter is one-half mile farther west. The altitude at the mill is approximately 3,000 feet.

ORE

Two principal types of ore are produced, a copper ore and a copper-zinc ore.

The principal sulfide minerals of the copper ore are bornite, chalcopyrite, pyrite, and some enargite. The ore occurs as large masses of sulfide in diabase, schist, quartzite, and shale in association with much introduced silica.

The copper-zinc ore contains principally chalcopyrite, sphalerite, pyrite, and small amounts of bornite. Some oxidized zinc minerals, such as hemimorphite (calamine), have been mined in small quantities. These minerals occur in a hard, siliceous gangue, and, as selective flotation of the copper requires grinding to about 70 or 75 percent minus 200-mesh, it is a very difficult ore to grind. Table 1 gives the average mill heads for 1942.

The following detailed description of the ore minerals is supplied by Dr. M. N. Short, professor of petrography, University of Arizona:

Pyrite (FeS_2)

The most abundant sulfide in the Magma mine is pyrite. Pyrite crystals are common in polished sections of the ore, but most of the pyrite is massive, without crystal faces. The pyrite grains almost invariably show embayments or veinlets filled by other sulfides. In almost every specimen quartz accompanies pyrite. In some specimens small grains of pyrite are attached to proportionately larger veins of quartz. In some tiny vugs, the space between acicular crystals of quartz is filled with quartz and sulfides.

Sphalerite (ZnS)

Sparse sphalerite occurs in the main ore bodies of the Magma mine as small grains surrounded by copper sulfides. Polished sections reveal galena embayments and veinlets, some of which are parallel to cleavage directions of sphalerite. The sphalerite tends to form cyrstal faces, and euhedral forms surrounded by other sulfides are common. Most of them show embayments occupied by galena or other sulfides.

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Enargite (Cu_3AsS_4)

Enargite occurs massive, with prominent, coarse cleavage. The cleavage surfaces generally show rounded areas of pyrite. In polished sections, enargite is almost invariably intergrown with tennantite.

Chalcopyrite ($CuFeS_2$)

The most abundant copper sulfide is chalcopyrite. Above the 3200 level it accounted for almost, if not quite, as much copper as bornite. In the low-grade pyritic ores, chalcopyrite is present in various amounts, whereas bornite is lacking or sparse. Some chalcopyrite occurs in most specimens of bornite.

Bornite ($CuFeS_4$)

The Magma mine has long been noted for its bornite ores. Thin sections of bornite ores show bornite containing unreplaced remnants of rock minerals, bornite embayments in calcite cleavage planes, and bornite veinlets cutting across sericite and penetrating between sericite plates. In rich sphalerite ores, bornite is almost absent, and the copper is represented by chalcopyrite or less commonly tennantite. Bornite cyrstals have not been observed in the Magma mine. In general, bornite has a relatively weak tendency to form crystal outlines, even in open spaces. Bornite intergrowths with chalcopyrite, with tennantite, and with chalcocite are common.

Galena (PbS)

In the Magma mine, galena is almost universally associated with sphalerite but is nowhere found in commercial quantities. Intergrowths of galena with bornite, and galena with chalcopyrite have been observed but are not important.

Chalcocite (Cu_2S)

Chalcocite never occurs in isolated masses more than 5 millimeters in diameter. Microscopic examination shows that it is everywhere intergrown with bornite. In general, the chalcocite is found only with the richer bornite ores, but not all rich bornite ores contain chalcocite.

Covellite (CuS)

The covellite is almost invariably associated with more or less malachite and mottled chalcocite. The amount of covellite in the ore is insignificant.

Because of the depth of the mine (now 4,800 feet) the ore is sometimes exposed to atmospheric action for long periods, and a film of oxidation products is formed which sometimes extends deeply into the minerals. This oxidation of

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the copper and iron sulfides sometimes makes flotation difficult, not only because of the presence of the oxidized material itself but also because the soluble sulfates interfere with flotation of the sulfides.

HISTORY OF MILLING

The history of milling operations from the beginning to 1930 is taken largely from an earlier publication.4/ The first section of the concentrator probably was completed in August 1914; the capacity was 150 tons a day. Both gravity and flotation processes were employed. Wilfley tables and deep Callow pneumatic cells were used. High-grade ore picked from the belt and the table and flotation concentrates were hauled to Magma over the Magma Arizona Railroad, which was then a narrow-gage road. At Magma the material was transferred to the standard-gage cars of the Arizona Eastern Railroad and hauled to the smelter at Hayden.

The flotation reagents first used were coal tar and creosote in an acid circuit. In 1915 the circuit was changed to alkaline, lime being employed as the alkaline reagent.

In 1915 a second section of 50 tons capacity was added to the mill to treat lead-zinc ore, but after 3 months of operation on this ore this section was changed over to treat copper ore. The addition of a tube mill to replace a small Hardinge mill increased the capacity to 150 tons.

In 1922 the flow sheet of these first two sections was rearranged, and a third section was added. These changes brought the total capacity to 600 tons.

The coal tar and creosote which had first been used as flotation reagents had been replaced by X-cake (alpha naphthylamine), and this in turn was replaced by a solution of thiocarbanalide in orthotoluidine (T-T mixture). Because of the scarcity and cost of the T-T mixture aerofloat 25 was adopted in 1928, and equally good concentration was obtained at a lower cost. The reagents now in use are named in the discussion of flotation.

After the smelter at Superior was completed in 1924, concentrates were transported to it by pumping instead of being shipped to the smelter at Hayden.

Before 1926 the ore was delivered to the mill by an aerial tramway operated during two 8-hour shifts. This method of delivery was replaced in 1926 by the present motor-haulage system and receiving bins. Installation of this latter system reduced the time required for ore delivery to one 8-hour shift, but ore is usually hauled during the second shift also and on Sundays when necessary.

In August 1928 a 4-foot Symons cone crusher was installed to replace two 24-inch Symons horizontal disk grinders previously used. However, one of these 24-inch grinders was retained as a stand-by for the 4-foot machine. This change reduced the crushing time for 800 tons of ore from 12 hours to 8, since the Blake jaw crusher, which had been overloaded, could be set at 3-1/2 inches instead of the former 2 inches.

4/ Work cited in footnote 3.

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In 1936 the deep Callow cells were replaced by three No. 36 Agitair machines. At first these machines had only eight cells, but four other cells were soon added to each set to increase recovery.

In early 1937 a fourth section of the mill was installed to treat copper-zinc ore. This section has a capacity of 450 to 500 tons and contains two No. 66 Marcy mills, which operate in closed circuit with two 60-inch Akins classifiers. With the addition of this section two more Agitair machines, similar to those already in use, were installed - one for selective flotation of the copper from the zinc and the other for the zinc rougher flotation. At the same time two six-cell No. 36 Agitair machines were installed as cleaners one to clean the rougher copper concentrate from the zinc ore and the other to clean the zinc rougher concentrate. This proved unsuccessful in both cases and both cleaners were applied to cleaning of the zinc rougher concentrate to raise it to satisfactory grade.

At about the same time a filter plant was installed to handle the cleaned zinc concentrate, using a 12- by 12-foot continuous vacuum filter, which was brought from the smelter.

In March 1943 this section of the mill was changed over to treat copper ore, while the copper-zinc ore was sent to section 1.

In December 1943 a six-cell No. 36 Agitair machine was installed to augment the rougher flotation of the zinc.

Data from operation of the mill for 1942 are given in tables 1 to 6.

GENERAL DESCRIPTION

The mill is on the western slope of a hogback, the southern end of which forms one side of the No. 2 tailing pond. The mill is of timber and corrugated iron construction except section 4, which is fireproof. Adequate protection against fire is provided by sprinkler heads and hydrants, except section 4, where only hand extinguishers are needed for protection against electrical fires.

Carpenter work and smaller machine work are done at the mill, while heavy machine work is done in the machine shop at the smelter.

The mill crew consists of about 50 men, 15 of whom (5 on each shift) are engaged in mill operation. Each operating crew consists of a shift boss, mill operator, flotation operator, oiler, and helper. The remaining men are distributed among the repair and shop crew, "bull gang," crusher crew, and zinc-filter-plant operators. All these men are under the supervision of a general foreman.

The fresh-water supply for the mill is obtained from the mine. The water flows by gravity from the mine tank on the 500 level (surface) through a 4-inch line to a storage pond at the foot of the western side of the mill.

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In case of insufficient supply, water is brought from the No. 2 tailing pond by an emergency siphon line. The water-storage tanks for mill feed are 129 feet above the surface of the pond. This fresh water is used for the zinc section only, while water for the copper sections is made up of reclaimed water from the two tailing ponds. A spillway on the northwest side of the No. 2 pond provides an intermittent flow to the lower No. 3 pond, and a spillway at the latter provides intermittent flow to the reservoir at the smelter. The upper pond has two centrifugal pumps, with a total capacity of 1,000 gallons a minute. The lower pond has two centrifugal pumps, with a total capacity of 600 gallons a minute.

Approximately 964 gallons of water is used for each ton of ore milled.

Tailing disposal is alternated between No. 2 and No. 3 ponds. This subject is discussed in greater detail under Disposal of Tailing.

ELECTRIC POWER

The electric power supplied to the mill is 25-cycle 3-phase alternating current, which is either generated at the smelter by waste-heat and direct-fired boilers or is obtained from the Salt River Valley Water Users' Association. Power is delivered to the mill at 2,200 volts, but it is transformed to 440 volts at the mill substation for all motors of less than 100 hp., except that which supplies air for the flotation machines.

ORE HAULAGE

The copper-zinc ore comes from the No. 2 shaft and the copper ore from the No. 3 and No. 5 shafts. It is hauled in gable-bottom cars of 3.5 tons capacity 42 tons per train of 12 cars. The track is laid with 30-pound rails and is of 18-inch gage. The trolley motor is supplied with current at 135 volts D. C. by a motor-generator set. The loaded cars are weighed by the train crew, six cars at a time, on the way to the mill on a track scale of 60,000 pounds capacity; empty cars are weighed on the return trip.

MILLING METHODS AND EQUIPMENT

Crushing, Grinding, and Tabling

In some respects the flow sheets of the four sections of the mill are similar, and in some respects different. In the following discussion these sections are described together wherever they are similar. The flow sheets are shown in figures 2, 3, and 4.

Crushing

Figure 1 is the flow sheet for crushing.

The mine ore, which is transported in cars, is dumped into a receiving bin of approximately 1,100 tons capacity. The east quarter of this bin is

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partitioned for the copper-zinc ore. The grizzly rails on top of the bin are spaced at 10 inches. The bottom slope of the bin is 45°, and the inside surface of the front is protected by worn-out ball-mill liners. The grizzly tender breaks up any large boulders and takes moisture samples.

Ore is drawn from the bins by eight 30-inch apron feeders. By the operation of one feeder at a time, lowering of the ore in the bins is kept even. The feeders are driven by a 5-hp., 100-volt, D. C., variable-speed motor of 500 to 1,500 r.p.m., the speed being controlled by the crusher operator. The speed of the apron feeders is 4 to 12 feet a minute. Three swing hammers over each feeder restrict the flow of ore.

The ore falls from the feeders onto a grizzly 15 by 24 inches in size, with 2-inch openings and sloping at 45° to break the fall of large boulders and prevent injury to the belt. Use of this grizzly also distributes the larger pieces of the ore on top of the smaller and thus makes hand picking easy.

Both oversize and undersize of this grizzly fall onto a 30-inch inclined conveyor belt. Near the head pully two men pick the ore, one removing tramp iron and wood chips and the other the high-grade material. This high-grade material is dropped into a car below the floor, which is picked up by the motor-haulage crew and taken to the mine for transportation to the smelter.

This 30-inch conveyor belt is 516 feet in length and is driven by a 15-hp., 440-volt, 713-r.p.m. motor at a belt speed of approximately 145 feet a minute. A magnetic head pulley energized by 25 amperes at 110 volts catches any tramp iron missed by the belt picker.

The ore drops from the conveyor onto a grizzly with 4-inch spaces and inclined at 45°. The oversize passes directly to a 12- by 24-inch Blake crusher set at approximately 3-1/2 inches. This crusher is driven at 300 r.p.m. by a 35-hp., 440-volt, 720-r.p.m. motor. The crushed product passes through an inclined chute to the 4-foot Symons cone crusher.

The undersize from the grizzly passes to a second grizzly 1 foot below the first, with 3/4-inch spacing. The oversize from this grizzly joins the product of the Blake crusher and passes to the Symons cone. The undersize drops onto a 24-inch conveyor belt, by which it is carried to the fine-ore bins.

The opening of the Symons cone crusher is set at approximately 5/8 inch for copper ore and 1/2 inch for copper-zinc ore. This crusher is driven by a 100-hp., 2,200-volt, 715-r.p.m. motor. One of the 24-inch Symons horizontal disk crushers which were replaced by the 4-foot cone crusher is retained as a stand-by for the latter. This horizontal disk crusher is driven by a 50-hp., 440-volt, 735-r.p.m. motor.

The product from the Symons cone crusher joins the undersize from the 3/4-inch grizzly and is discharged onto a 24-inch inclined conveyor belt 125 feet in length and traveling at approximately 150 feet a minute. This belt discharges either directly into the copper-zinc fine-ore bin or onto an 18-inch

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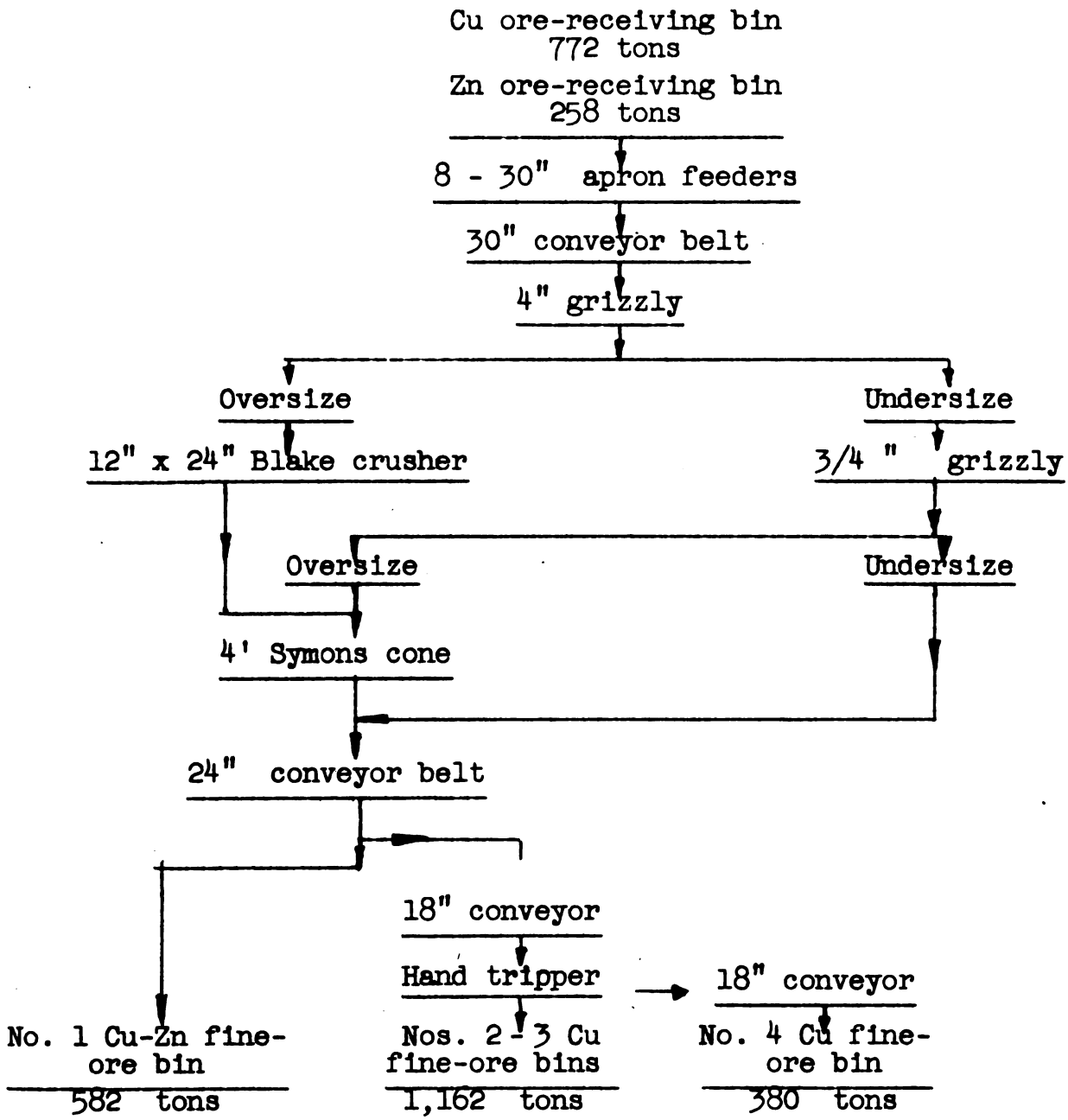

FIGURE 1.- Flow sheet of crushing plant.

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cross conveyor belt with a hand tripper. By means of this tripper the ore is discharged either into the fine-ore bins of sections 2 and 3 or onto another 18-inch conveyor belt which carries it to the fine-ore bin of section 4.

The zinc fine-ore bin and the copper ore bins of sections 2 and 3 have flat bottoms. The capacity of the former is 582 tons and that of the latter, 1,162 tons. The fine-ore bin of section 4 is a cylinder 17 feet in diameter and 29.08 feet in height, with a capacity of 380 tons.

A fan collects the dust from both crushers and discharges it into the copper-zinc fine-ore bin. This is driven by a 10-hp., 440-volt, 1,490-r.p.m. motor.

Coarse Grinding, Sections 1, 2, and 3

The flow sheets for treatment subsequent to crushing are shown in figures 2, 3, and 4.

The coarse-grinding flow sheets of section 1 (copper-zinc ore) and section 2 and 3 (copper ore) are identical, except that tables are used in sections 2 and 3. Ore is drawn from the fine-ore bins by 24-inch pan feeders driven in sections 2 and 3 by the same motors that drive the screens, drag classifiers, and elevators and in section 1 by a separate motor. The quantity of ore carried by the feeders is controlled by a gate with a hand wheel, which regulates the depth of the stream of ore.

The coarse-grinding mills are 64-1/2 Marcy mills driven in sections 1 and 2 by 100-hp., 2,200-volt, 725-r.p.m. motors and in section 3 by a 150-hp., 2,200-volt, 725-r.p.m. motor. The same motors drive the fine-grinding tube mills through line shafts. The Marcy mills are driven at 25 r.p.m.

The grinding medium in the copper sections is 5-inch forged chrome-steel balls. The ball consumption is 1.15 pounds a ton in the copper sections and 0.45 pound a ton in the copper-zinc section. A 4-inch ball of the same type is also used on the zinc section, with a consumption of 0.68 pound a ton. The ball load is 10,000 pounds, and the pulp density is maintained at approximately 75 to 78 percent solids. Chilled cast-iron liners are used on the feed ends of the mills, with 6-inch manganese-steel liners for the remainder. The grates are of chrome steel, with 3/16-inch openings. The breast liners last for about 1 year, feed-end liners about 4 months, and grates about 6 months.

Coarse Grinding, Section 4

The ore is drawn from the fine-ore bin by 24-inch pan feeders on 6-inch rollers, the quantity of feed being controlled by rack and pinion gates. The ore is carried by a 24-inch cross-conveyor belt driven by a 2-hp., 440-volt, 710-r.p.m. motor to a vertical chute through which it passes to the Marcy mill. In this section both the primary and secondary mills are No. 66 Marcy mills, each driven by a 125-hp., 2,200-volt, 730-r.p.m. motor with texrope drive.

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These mills, like those in the other sections, rotate at 25 r.p.m. and the same types of liners are used, but the normal ball load is 11,000 pounds. Pulp densities in the mills are maintained at 75 to 78 percent solids.

Fine Grinding, Screening, Classification, and Tabling

The flow sheets show that in sections 1 and 4 all final concentration is done by flotation, while in sections 2 and 3 a portion is done by tables. In sections 1, 2, and 3 the product of the primary Marcy mill is separated by 14-mesh screens, and in section 4 this separation is made by a classifier. In each of these sections the oversize goes back to the Marcy mill.

In section 1 the undersize of the screens goes to a drag classifier, from which the sands are sent to a tube mill, and the product goes back to the classifier. All of this material goes to flotation. In sections 2 and 3, the undersize of the screens goes to tables. The concentrate from the tables, after thickening, goes to the smelter, while the tailing, after regrinding of the coarser material, goes to flotation. In section 4, the product of the Marcy mill goes to a classifier, from which the sand is returned to the mill; the overflow, after regrinding of the coarser part in a second Marcy mill, goes to flotation.

Elevating, Sections 1, 2, and 3. - Elevating is necessary in two places in each of these sections. In each section one elevator handles the discharge from the Marcy mill. This contains 75 to 78 percent solids as it leaves the mill, but water is added in the discharge boxes in sufficient volume to reduce the content of solids to 45 or 50 percent. This material flows to the elevator sumps.

In section 1 the second elevator handles the undersize of the Callow screens and the discharge of the tube mill. In sections 2 and 3 the second elevator handles the table tailing and the discharge of the mill.

The elevator shafts are of wood. The head pullys are 42 inches in diameter and the tail pullys 30 inches. The distance between pullys is 40 feet. The belts are 16 inches in width, with 7- by 14- by 6-inch buckets spaced at 15-inch intervals. The belt speed is 540 feet a minute, and the pulp load is approximately 45 tons an hour. In each section the elevators, screens, and drag classifiers are driven by a 20-hp., 440-volt, 730-r.p.m. motor.

Screening, Sections 1, 2, and 3. - In each section the discharge from the elevator which raises the pulp from the Marcy mills passes by a launder to two 24-inch, 14-mesh Callow screens, between which the feed is divided equally. Additional water for screening is supplied by sprays. The screen speed is 110 feet a minute, and the removal of minus 14-mesh material is approximately 70 percent. The oversize of the screens passes by gravity to the scoop boxes of the Marcy mills. This oversize is approximately 45 percent of the original feed to the mills.

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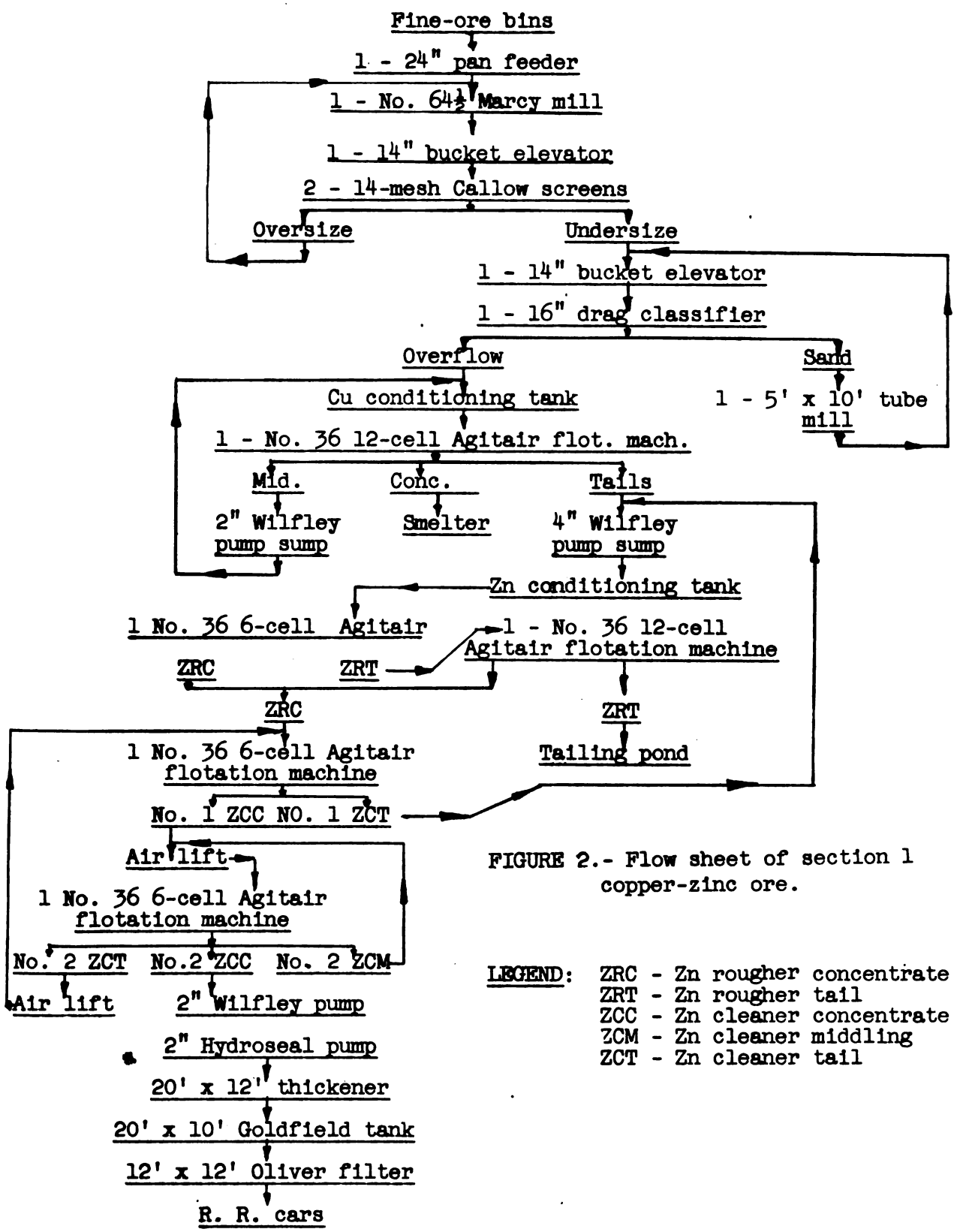

FIGURE 2.- Flow sheet of section 1 copper-zinc ore.

LEGEND:
ZRC - Zn rougher concentrate
ZRT - Zn rougher tail
ZCC - Zn cleaner concentrate
ZCM - Zn cleaner middling
ZCT - Zn cleaner tail

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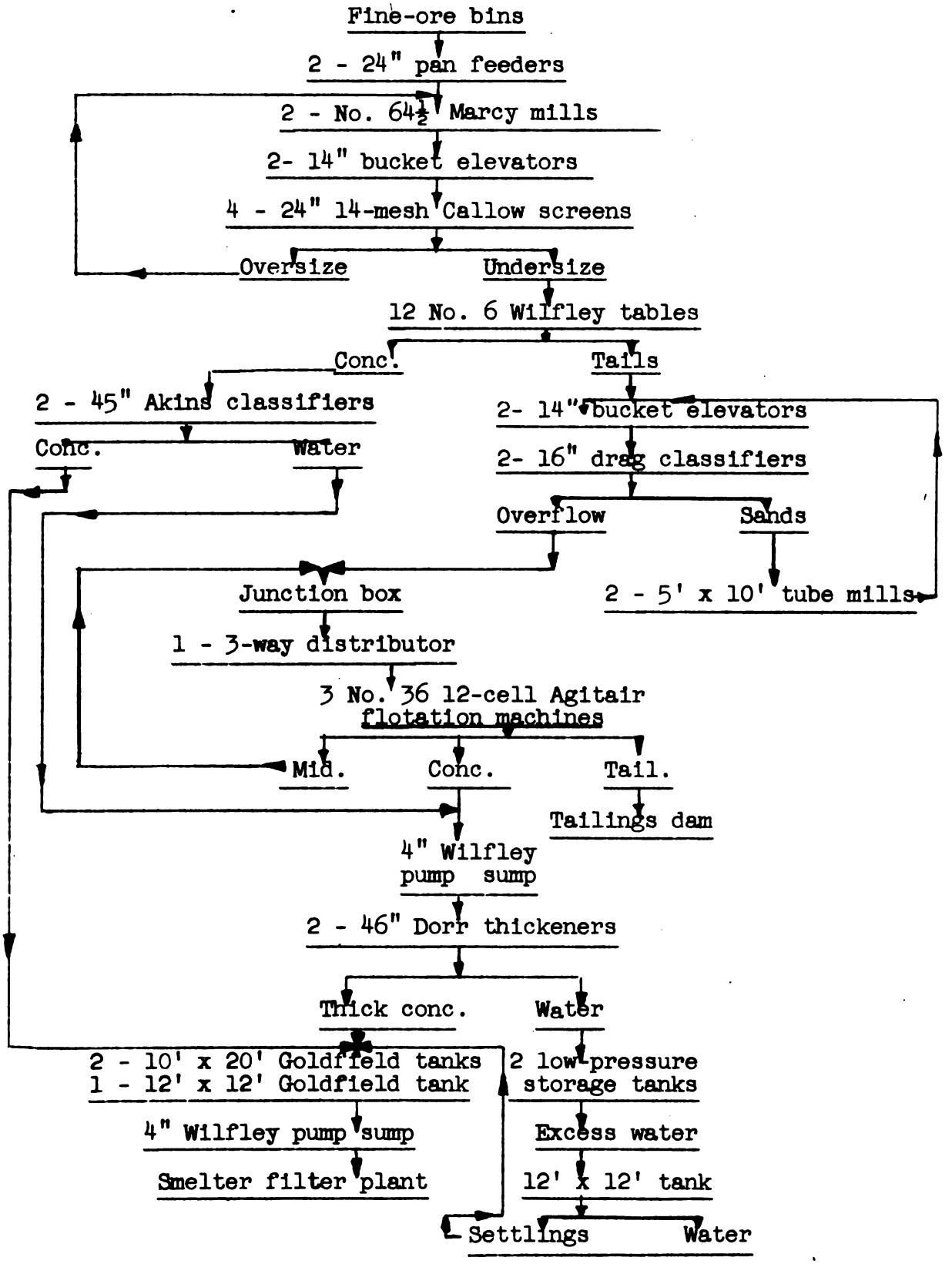

FIGURE 3.- Flow sheet of sections 2 and 3, copper ore.

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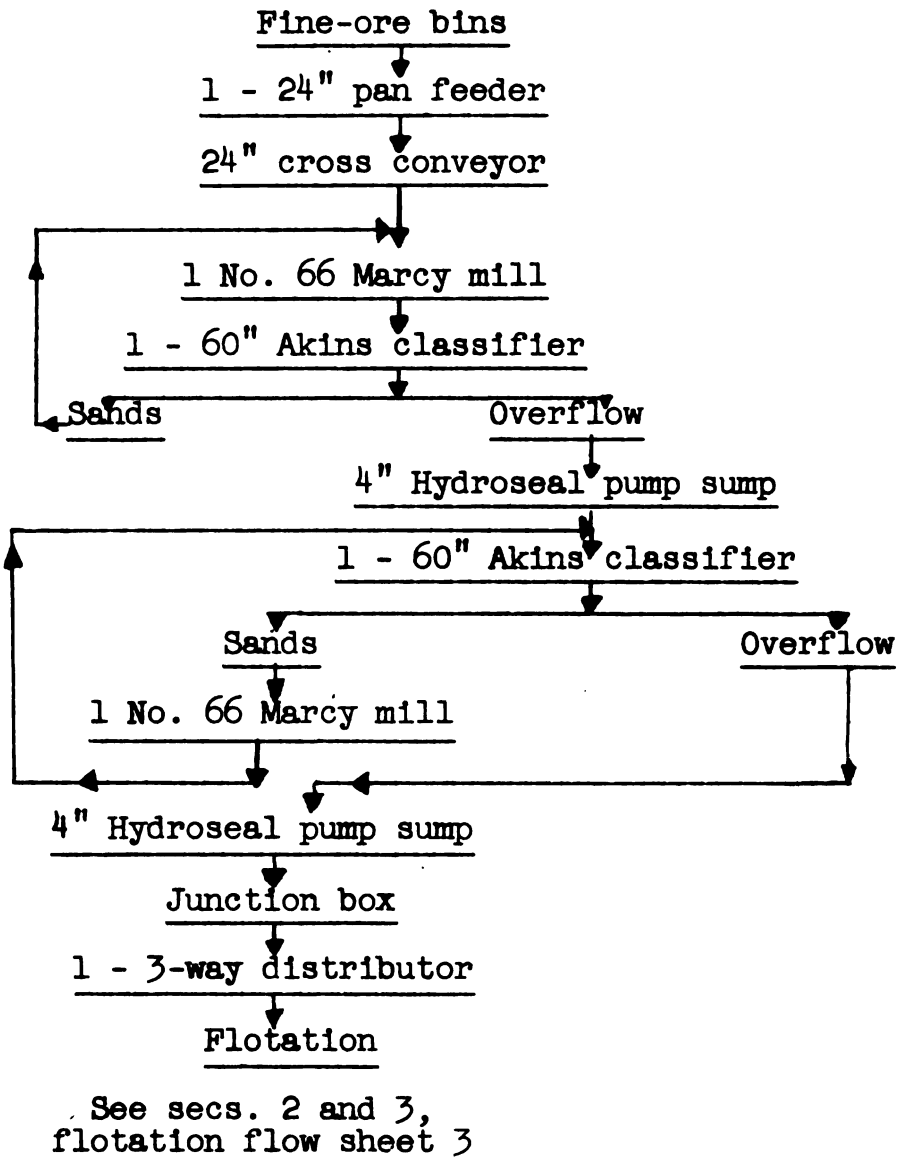

FIGURE 4.- Flow sheet of section 4, copper ore.

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Classification and Fine Grinding, Section 1. - The undersize of the screens flows to the sump of an elevator by which it is lifted to a drag classifier 2-1/2 feet in width and 2 feet in depth, extending horizontally for 23 feet and then sloping upward at an angle of 20° for 15 feet. The drag is constructed of 16-inch elevator belting to which 1/4- by 2-inch angle irons 8 inches in length are bolted in staggered position at 8-inch intervals. The head pully is 42 inches in diameter. The tail pully and the idler pully at the bottom of the slope are 31 inches in diameter. The bottom of the slope is lined with 18- by 18-inch steel plates 1-1/2 inches in thickness. The speed of the belt is 130 feet a minute. The feed enters at a point 4 feet up the slope. The fine material overflows through 4-inch by 4-1/2-foot weirs cut in the sides near the end of the horizontal section. Enough water is added to the feed to make the density of the overflow approximately 20 percent solids.

The sand from the classifier passes by a launder to a tube mill 5 by 10 feet in size, with trunnion discharge. The liners are chilled cast iron except the lifter bars, which are manganese steel. The grinding medium is 2-inch chilled cast-iron balls, and the load is 18,000 pounds. The mill is driven by the same motor that drives the Marcy mill.

The discharge of this mill goes back to the drag classifier; thus all feed to the section eventually passes the overflow of this classifier, and its fineness is determined by the rate of overflow. This material goes to flotation with a density of 20 to 25 percent solids.

Classification, Tabling, and Fine Grinding, Sections 2 and 3. - The undersize from the Callow screens is carried by a launder to 12 No. 6 Wilfley tables, 6 for each section. Feed is distributed by wooden splitters and passes to the tables through 4-inch vertical pipes. A 4-inch low-pressure water line serves all the tables, with a 1-inch line feeding the water box of each table.

Each group of six tables is driven by a 10-hp., 440-volt, 715-r.p.m. motor at 240 strokes a minute, with a throw of 3/4 inch.

The tables are covered with 1/8-inch linoleum. Accumulation of scale from the lime in the feed water gives considerable trouble, and the covers have to be replaced after 8 to 10 months use.

Only two products, concentrate and tailing, are made by the tables. The concentrate is a finished product. A recovery of 26 to 28 percent of the copper in the feed is obtained at a concentration ratio of 7.80 to 8.10 : 1. Analyses of feed and products are given in table 1. This concentrate, containing 15 to 18 percent solids, flows by gravity to two 45-inch by 13-foot simplex low-weir Akins classifiers for dewatering. A third similar classifier is used for dewatering spillage and other clean-up material. These classifiers are driven at 3-1/2 r.p.m. by a 15-hp., 440-volt, 475-r.p.m. motor. The slope is 12°.

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The dewatered concentrate, containing 10 to 12 percent moisture, drops into a launder, where it is joined and carried off by the underflow of two Dorr thickeners which handle the copper flotation concentrate. This concentrate contains 65 to 68 percent solids.

The table tailing, which amounts to approximately 90 percent of the feed, goes to elevators, by which it is raised to drag classifiers similar to that in section 1.

The sand from the classifiers passes by launders to two tube mills similar to that in section 1. The charge of 2-inch balls is the same as in section 1, but some 3-inch balls are added. The consumption of 2-inch balls is 1.71 pounds a ton of ore.

The product of these mills returns to the classifiers; thus, as in section 1, the fineness of the material which goes to flotation is determined by the overflow of the classifiers.

Classification and Fine Grinding, Section 4. - The Marcy mill, which is fed from the fine-ore bin, is in closed circuit with a 60-inch double-pitch simplex high-weir Akins classifier. Water is added at the mill discharge box. The classifier is driven by a texrope drive from a 5-hp., 440-volt, 715-r.p.m. motor. A 3/4-hp. motor is used to raise the spiral when the section is shut down. The circulating load on the ball mill is approximately 35 percent of the original feed. The classifier overflow contains 50 percent solids, of which 40 percent is plus 65-mesh. This material flows to a sump in the basement. A chip remover on the overflow lip catches all tramp wood.

The material is pumped from this sump by a 4-inch Hydroseal pump driven by a 10-hp., 440-volt, 1,432-r.p.m. motor to the secondary Akins classifier. The density in this classifier is held at 22 to 25 percent solids. The overflow contains approximately 10 percent plus 65-mesh and 45 to 50 percent minus 200-mesh. This material flows to a second sump from which it is pumped, by a pump similar to that mentioned above and with similar drive, to a launder 30 feet above the sump, in which it flows to flotation.

The sand from the secondary classifier goes to the secondary Marcy mill, of the same type as the primary mill but charged with 2-inch forged chrome-steel balls. The ball load is 12,000 pounds. The consumption of 2-inch balls is 1.71 pounds a ton. A small proportion of 3-inch cast-iron balls is also used. The consumption of 3-inch balls is reported as 0.17 pound a ton, but this figure cannot be considered reliable, as it is derived only from the fact that a given weight of 3-inch balls is added each day without regard to the tonnage ground during the preceding day.

The consumption of 2-inch balls is given as 1.71 pounds a ton in each of the copper-ore sections. This is only an approximation because it is based on the total weight of balls added and the total tonnage of copper ore ground. In each section the weight of balls added each day is based on the quantity of ore ground in the preceding day, but the only permanent record kept is of the totals for the three sections.

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The circulating load on the secondary mill is approximately 45 percent, and the density is 75 to 78 percent solids. The discharge from this mill returns to the secondary classifier. Water is added to the discharge of the mill in sufficient quantity to obtain the desired density of classifier overflow, which is 22 to 25 percent solids.

Flotation

The flotation of the copper-zinc ore involves separation of the copper contained as well as separation of zinc from tailing. For this reason the flow sheet of section 1, in which this ore is treated, is more complicated than that used in the other sections, in which the only object is concentration of copper.

Copper-Zinc Ore, Section 1

Flotation of Copper. - The flotation feed is the overflow of the drag classifier. This material has been finely ground and contains 1.5 percent plus 65-mesh and 70 percent minus 200-mesh.

This feed flows by gravity to the copper conditioner, which is a tank 8 by 12 feet in size, with rakes driven by a 3-hp., 440-volt, 1,364-r.p.m. motor. The conditioning period for operation of the zinc depressants is approximately 25 minutes.

The discharge from the conditioner flows by gravity to the first 12-cell, No. 36 Agitair flotation machine. This machine makes three products - rougher copper concentrate, middling, and tailing.

The concentrate is taken from the first six cells. It contains 7 to 9 percent copper and goes to the smelter. A typical analysis of the copper concentrate is given in table 1.

The last six cells produce a copper middling which flows by gravity to a 2-inch Wilfley pump, driven by a 10-hp., 440-volt, 1,450-r.p.m. motor, by which it is returned to the copper conditioning tank.

The recovery of copper is 75 percent, and the concentration ratio is 7.5 to 1. The average loss of zinc in the copper concentrate is approximately 16 percent.

An approximately neutral circuit is maintained, the pH varying from 6.9 to 7.3.

The zinc depressants are zinc sulfate, sodium cyanide, and sodium sulfide. All are added to the Marcy-mill scoop box. The quantities per ton of ore are:

	Pounds
Zinc sulfate	1.75 to 2.0
Sodium cyanide	.025
Sodium sulfide	1.0 to 1.2

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The main copper promoter is thiocarbanalide (T-125). This is added to the drag classifier sand between the classifier and the tube mill. The quantity used is 0.05 to 0.08 pound a ton of ore. Potassium ethyl xanthate (Z-3) is frequently added in the launder between the copper conditioner and the flotation machine, the quantity being approximately 0.01 pound a ton of ore. Pine oil, when needed, is added to the copper middling in the flotation machine, the amount varying from 0.006 to 0.010 pound a ton of ore. Aerofloat 242 is used occasionally for its ability to collect precious metals. This is added with the thiocarbanalide between the drag classifier and the tube mill. The amount used does not exceed 0.025 pound a ton of ore.

Flotation of Zinc. - The tailing from the copper flotation contains approximately 84 percent of the original zinc. It flows by gravity to a sump in which 1.2 to 1.4 pounds of copper sulfate is added to reactivate zinc per ton of total copper-zinc ore milled.

This material is pumped to the zinc conditioner by a 4-inch Wilfley pump driven by a 50-hp., 440-volt, 720-r.p.m. motor. The conditioner is the same size as that used for conditioning before flotation of the copper and, like it allows a conditioning period of approximately 25 minutes.

From the conditioner the material flows by gravity to a six-cell No. 36 Agitair machine. This makes only two products, a rougher concentrate and a primary rougher tailing, neither of which is a finished commodity. The concentrate flows by gravity through a launder to the first cleaner, and the tailing flows through an 8-inch pipe to a 12-cell No. 36 Agitair machine, in which the last of the zinc is removed. This machine produces a rougher concentrate from all cells. This joins the concentrate from the first zinc machine and goes to the first cleaner. The tailing, which is a finished product, goes to a tailing pond.

The combined concentrate from the 6-cell and 12-cell machines contains approximately 25 percent zinc, 20 percent iron, and 15 to 20 percent insoluble material. This material flows by gravity to the first cleaner, which is a 6-cell No. 36 Agitair machine. All 6 cells of this machine produce the primary zinc cleaner concentrate. The tailing flows by gravity to join the tailing from selective copper flotation. It is then pumped back to the zinc conditioner.

The primary zinc cleaner concentrate contains approximately 35 percent zinc and 12 percent iron. It flows by gravity to the sump of a 3-inch air lift, by which it is raised to the second cleaner, at the same elevation as the first, and, like it, having six cells. The first three cells of this machine produce the final zinc concentrate, which contains approximately 50 percent zinc, 4 to 7 percent iron, and 2 to 3 percent insoluble material. This flows by gravity to a sump under the machine, from which it is pumped to the zinc thickener by a 2-inch Wilfley pump and a 2-inch Hydroseal pump, used either singly or together. Each is driven by a 5-hp., 440-volt, 1,450-r.p.m. motor.

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The last three cells of this machine produce a middling which flows by gravity to the air lift which raises the concentrate product of the first cleaner. It is thus returned for re-treatment in the same machine in which it was produced.

The tailing from the second cleaner flows by gravity to another 3-inch air lift which raises it to the first cleaner.

The tailing from the copper concentrate is practically neutral. It is made alkaline by the addition of milk of lime to the launder in which the rougher concentrate flows to the No. 1 cleaner. Alkalinity of the rougher cells is created and maintained by the tailing from the No. 1 cleaner. The alkalinity of the No. 1 cleaner is usually kept at the equivalent of about 0.50 pound of CaO a ton of water, while that of the No. 2 cleaner is about 0.35 pound. If higher alkalinity is wanted in the second cleaner, milk of lime is fed to the concentrate from the first cleaner. The alkalinity of the rougher cells, which is given to them by the tailing from the first cleaner, is the equivalent of approximately 0.22 pound of CaO a ton of water, and the pH value is approximately 10.5 to 10.8. The quantity of dry lime used varies from 6.0 to 6.2 pounds a ton of total ore milled.

The promoter used for zinc is potassium amyl xanthate (Z-5); 0.10 pound a ton of ore is added in the launder between the zinc conditioner and the flotation machine. An additional 0.10 to 0.18 pound a ton is stage-fed to the rougher cells. Pine oil, when needed, is added in the zinc conditioner; the quantities range from 0.10 to 0.20 pound a ton. All weights of reagents are based on the total tonnage of ore treated in section 1 and not on the tonnage of tailing from the copper rougher.

Thickening and Handling Zinc Concentrate. - The zinc concentrate is thickened in a 12- by 20-foot Dorr thickener on the eastern side of the ridge above the zinc filter plant. The rakes make one revolution in 6 minutes and are driven by a 1-1/2-hp., 440-volt, 1,450-r.p.m. motor. The overflow water runs by gravity to the low-pressure storage tanks. The underflow, containing 70 to 72 percent solids, flows by gravity to a 10- by 20-foot Goldfield storage tank. The rakes are driven at a speed of 5 r.p.m. by a 5-hp., 440-volt, 715-r.p.m. motor. The discharge from this tank, containing 70 to 72 percent solids, flows by gravity to a 12- by 12-foot continuous vacuum filter. The vacuum pump is driven by a 25-hp., 440-volt, 715-r.p.m. motor. The filter is driven by a 5-hp., 440-volt, 1,450-r.p.m. motor. The filtered concentrate, which contains 10 to 12 percent moisture, is carried by a 24-inch conveyor belt, from which it is discharged into railroad cars for shipment to the smelter at El Paso, Tex. The filter water is pumped back to the Dorr thickener.

One man operates the filter, and a helper distributes the concentrate in the car. A Stephens-Adamson car loader is used to distribute the concentrate when box cars are used.

Copper Ore, Sections 2, 3, and 4

Copper Flotation, Sections 2, 3, and 4. - The flotation feed is the reground table tailing in sections 2 and 3 and the overflow of the second classifier in section 4. This material flows to a junction box about 20 feet

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above the flotation cells. From this, it passes to a three-way distributor which distributes the feed equally by means of three weirs. If one cell is shut down, the feed is cut off by filling in the proper weir. The feed contains 18 to 22 percent solids, which consist of approximately 8 percent plus 65-mesh and 45 to 50 percent minus 200-mesh. Typical analyses of the flotation feed, concentrate, and tailing are given in table 1.

Flotation is accomplished in three 12-cell No. 36 Agitair flotation machines.

Air for the flotation cells of all sections is supplied by a No. 11 Sturtevant blower which operates at 198 r.p.m. and has a capacity of 5,200 cubic feet a minute. Air is delivered to the flotation cells at a pressure of 1.5 pounds a square inch. Excess air which is not used in the cells is bypassed automatically. The blower is driven by a 75-hp., 2,200-volt, 720-r.p.m. motor. When only two sections of the mill are in operation, air is supplied by a No. 3 Roots blower of 1,600 cubic feet capacity, driven at 428 r.p.m. by a 10-hp., 440-volt, 715-r.p.m. motor.

The first six cells of each machine produce a concentrate which contains approximately 14 percent copper and which, after thickening, is sent to the smelter.

The last six cells of each machine produce a middling which contains 2 to 3 percent copper. This flows by gravity to a sump from which it is pumped by a 2-inch Wilfley pump driven by a 10-hp., 440-volt, 1,450-r.p.m. motor back to the junction box, where it joins the new feed.

The tailings from the three machines come together in a launder and flow by gravity either to the No. 3 tailing pond or to a tailing sump under the flotation floor, from which they are pumped to a launder 60 feet above the sump, in which they flow to the No. 2 tailing pond. A 4-inch Wilfley pump driven by a 40-hp., 440-volt, 720-r.p.m. motor and a 4-inch Hydroseal pump driven by a 25-hp., 440-volt, 750-r.p.m. motor are used alternately to pump the tailings.

A highly alkaline circuit is maintained by feeding milk of lime into the scoop boxes of the Marcy mills. The alkalinity is maintained at the equivalent of 0.30 to 0.35 pound of CaO a ton of water or a pH of 10.8 to 11.2 in the flotation cells. This requires approximately 3.9 pounds of lime a ton of ore. The lime is drawn from the mixing tank into a storage tank, from which it flows by gravity through a 2-inch line to a 2-inch centrifugal pump driven by a 3-hp., 440-volt, 710-r.p.m. motor. The pump delivers it to a 2-inch line running behind the Marcy mills. The volume entering each mill is controlled by a valve which is manipulated by the operator. The operator tests the alkalinity in the flotation cells every half hour from samples obtained from siphon filters in the feed launders just ahead of the junction box. Alkalinity is determined by titration with sulfuric acid. As the water samples are taken from the flow from each of the three sections of the mill, the operator can determine the alkalinity of each section and can adjust the valves controlling the lime delivery to each Marcy mill accordingly.

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The principal promoter is potassium ethyl xanthate (Z-3). This is added to the junction box in amounts of 0.17 to 0.19 pound a ton. Minarec B is used as a secondary promoter. In sections 2 and 3 this is added to the sand from the drag classifiers as it goes to the tube mills, and in section 4 it is added to the scoop box of the second Marcy mill. The quantity used is approximately 0.05 pound a ton. Pine oil is added to the junction box, the amount depending on the condition of the froth and ranging from 0.10 to 0.15 pound a ton, the larger figure being the more common.

The recovery in these three sections is 96 percent of the copper, 75 percent of the iron, 100 percent of the gold, and 88 percent of the silver in the flotation heads, with a concentration ratio of 3.5 : 1.

Thickening and Handling Copper Concentrate. - The copper flotation concentrate from sections 2, 3, and 4 is joined by the selective copper flotation concentrate from section 1, and all flows to a concrete sump 6 feet in length, 3 feet 6 inches in width, and 5 feet in depth. The concentrate contains 28 to 30 percent solids, but it is diluted in the sump to approximately 15 percent solids by water from the classifiers, which dewater the table concentrates of sections 2 and 3. Water from the clean-up classifier, which will be mentioned later, also enters at this point.

The concentrate is pumped to a distributing box approximately 80 feet above the sump by one of two 4-inch Wilfley pumps. One pump is driven by a 50-hp., 440-volt, 735-r.p.m. motor and the other by a 40-hp., 440-volt, 480-r.p.m. motor. The pumps are operated at 1,350 r.p.m. Runners last for about 60 days and cases for about 120 days.

The concentrate and water flow from the distributing box to two Dorr thickeners, each 9 feet 5 inches in depth and 45 feet 6 inches in inside diameter. The arms make one revolution in 10 minutes. They are driven by 3-hp., 440-volt, 750-r.p.m. motors, through reducing gears with ratio of 36.47 to 1, belts, shafts, and worm gears.

The overflow water runs to two 2,200-gallon supply tanks placed between the thickeners. These tanks supply water for the Wilfley tables, Callow screens, and drag classifiers of sections 2 and 3. Any excess water flows to a 12- by 12-foot tank, in which any remaining solids settle out. The overflow from this tank is piped to the tailing pond. The solids are washed out periodically with the minimum amount of water and sent through a 4-inch pipe to one of the Goldfield concentrate storage tanks.

The thickened flotation concentrate and the dewatered table concentrate, containing 68 to 75 percent solids, are discharged by gravity into a 10- by 20-foot Goldfield storage tank. The rakes are driven at 5 r.p.m. by a 5-hp., 440-volt, 715-r.p.m. motor. Additional storage is provided by a second 10- by 20-foot Goldfield tank, the rakes of which are driven by a 10-hp., 440-volt, 715-r.p.m. motor, and a third 12- by 12-foot Goldfield tank, the rakes of which are driven by a 5-hp., 440-volt, 715-r.p.m. motor. Continuous agitation is necessary to prevent accumulation of solids that might flow into the pump and choke it.

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Concentrate is transferred from the first tank to the second or third by allowing it to flow to a sump from which it is raised by the equipment used to pump concentrate to the filter plant.

From these tanks the concentrate is pumped twice a day (at 8 a.m. and 3 p.m.) to the filter plant at the smelter. The pipe line is 2,900 feet in length and 4 inches in diameter. It follows the general contour of the ground, rising to 92 feet above the pump and then falling 145 feet to the Goldfield tanks at the filter plant. A second 4-inch line parallels the first for use whenever the latter becomes choked or leaks. Most of the abrasion on the pipe is along the bottom and is caused by the table concentrate which settles to this part of the pipe.

The pipe is cleaned out both before and after pumping by the admission of air at 90 pounds pressure or water at 50 pounds pressure at the highest point of the pipe. Little trouble is experienced as long as the density of the material is at least 65 percent solids.

The concentrate is pumped by a 4-inch Wilfley pump driven at 1,600 r.p.m by a 100-hp., 440-volt, 725-r.p.m. motor. A stand-by motor of 75 hp., 440 volts, and 480 r.p.m. is provided. The pressure at the pump is 155 pounds a square inch. Pump runners last for 50 to 75 hours of pumping, follower plates for 20 to 30 hours, and casings for about 100 hours.

Mill Clean-Up

Drainings from the flotation machines and any other mill spillage flow by gravity to any one of four large sumps at the foot of the west side of the mill adjacent to the storage pond of the zinc section. The clear water passes on into the storage pond. The settled material is pumped out by a 2-inch Hydroseal pump driven by a 7-hp., 440-volt, 1,450-r.p.m. motor, which is mounted on a truck and moved from sump to sump as needed. The material is diluted with high-pressure water to prevent choking the pump. This material is pumped through a 2-inch line running up the side of the mill into a 45-inch by 13-foot Akins classifier similar to those used for dewatering the table concentrate.

Water and solids are disposed of in the same way as those from the table concentrate, which is treated in similar classifiers. The water flows to the concentrate sump ahead of the thickeners, and the solids drop into a launder and are carried away by the thickened concentrate from the two Dorr thickeners.

Sampling

Tonnage Samples

In sections 1, 2, and 3 tonnage samples are cut by automatic samplers as the ore drops from the fine-ore feeders to the scoop boxes of the Marcy mills. In section 1 the sampler is similar to an elevator bucket and is 3-1/2 by 5-3/4 by 8-1/2 inches in size. It is attached by an iron strap to a 6-inch horizontal belt, and its weight is supported by a small wheel running on a track. The bucket passes through the full width of the ore stream. It is driven indirectly from the ore feeder, and a sample is taken and dumped into a sample box every 2-1/2 minutes.

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In sections 2 and 3 the samplers are rectangular iron chutes 4-1/2 by 7-1/2 inches in size, which are passed across the ore stream by the piston of a horizontal water cylinder 4 inches in diameter with 21-inch stroke. A weighted lever attached to a crank arm on the old dry-lime feeder controls the flow of water. Samples are cut every 5 minutes.

The contents of the sample boxes are weighed every 2 hours, and the sums of the four net weights obtained in each section at the end of each shift are placed on the operations sheet.

In section 4 the tonnage sample is taken by stopping the conveyor belt to the primary Marcy mill and scraping off all of the ore on a section of the belt measured with a 10-foot pole. Only three such samples are taken during a shift.

Mill tonnages determined in this manner check the tonnages of ore weighed by the haulage crew on the track scales within 1 percent.

Mill Samples

In sections 1, 2, and 3 the head samples are cut from the undersize of the Callow screens by automatic samplers operated by water tipping boxes. In section 4 the head sample is cut from the overflow of the secondary classifier by an automatic sampler. Water tipping boxes are also used to operate samplers on table concentrate, table tailing, copper flotation concentrate, selective copper flotation concentrate, zinc cleaner concentrate, copper and zinc flotation tailing, and over-all mill tailing. The zinc rougher concentrate is sampled by hand every half hour.

Disposal of Tailing

The final tailing from the flotation machines, consisting of the tailing from copper flotation in sections 2, 3, and 4 and the tailing from the zinc rougher in section 1, is discharged into a launder 10 by 12 inches in size, with a slope of 3/16 inch a foot. This tailing, which contains 18 to 20 percent solids, flows by gravity to the No. 3 tailing pond south of the mill and approximately 250 feet below it.

Holes 1-1/4 inches in diameter and spaced 10 feet apart in the bottom of the launder discharge the tailing into 3- by 4-inch launders, which deposit it 3 or 4 feet from the bank. Approximately 200 feet of discharge launder is required to deposit the tailing from all sections.

The clear water is removed by the suction pipes of two reclaiming pumps, both four-stage centrifugal pumps rated at 300 gallons a minute with a 500-foot head. They are driven at 1,400 r.p.m. by 75-hp., 440-volt, 725-r.p.m. motors. These pumps lift the water to two 16,000-gallon supply tanks on a level with the top of the mill. Approximately 75 percent of the water is recovered from the tailing pond, and the remainder passes over a spillway and flows to a reservoir near the smelter.

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As a general rule, tailing is sent to the No. 3 pond during the day shift only. During the other two shifts the over-all tailing flows by gravity into a sump 9 feet in length, 7 feet in width, and 6 feet in depth under the flotation floor. The material is pumped from this sump by either a 4-inch Wilfley pump driven by a 40-hp., 440-volt, 720-r.p.m. motor or a 4-inch Hydroseal pump driven by a 20-hp., 440-volt, 1,450-r.p.m. motor. These pumps lift the tailing into a discharge box about 50 feet above the pumps, from which it flows by gravity to the No. 2 pond, where it is distributed in the same way as at the No. 3 pond.

The slopes of the banks of both dams are 28°. A spillway in the No. 2 dam permits excess water from sudden rain to escape, and any water that passes this spillway flows to the No. 3 pond.

The clear water from No. 2 pond goes to two centrifugal reclaiming pumps with capacity of 500 gallons a minute, driven by 30-hp., 440-volt, 1,450-r.p.m. motors. These pumps raise the reclaimed water to the two 16,000-gallon tanks mentioned above.

This reclaimed water is used only in sections 2, 3, and 4, the copper sections. Water for section 1, the zinc section, is obtained from the mine. The pH of the mine water varies from 6.8 to 8.0.

Delivery of this mine water to the zinc-ore storage pond is described on page 6. The water is pumped from this pond by one of three Aldrich triplex pumps to a 16,000-gallon supply tank 129 feet above the pond. These pumps are a 5-1/2 by 9-inch pump, rated at 200 gallons a minute against a 300-foot head, driven by a 25-hp., 440-volt, 725-r.p.m. motor; a 6- by 9-inch pump, rated at 200 gallons a minute against a 500-foot head, driven by a 35-hp., 440-volt, 485-r.p.m. motor; and a 7- by 7-inch pump, rated at 200 gallons a minute against a 200-foot head, driven by a 10-hp., 440-volt, 1,450-r.p.m. motor.

SMELTING METHODS AND EQUIPMENT

The complete flow sheet of the smelting plant is given in figure 5. Data on smelter operations are given in tables 7 to 14.

The construction of the smelter was begun in February 1923 and completed in April 1924. The original capacity was approximately 500 tons of dry charge in 24 hours. By the addition of a sixth roaster in 1930 the capacity was increased to 600 to 650 tons a day.

The construction of a natural-gas pipe line to Superior was completed in January 1936, and thereafter the reverberatory furnace, roasters, direct-fired boilers, and assay furnaces were fired with gas.

Auxiliary Equipment and Labor

Smelter Stack

The stack is 300 feet in height. The inside diameter at the top is 20 feet. Approximately 250,000 cubic feet of gas is discharged a minute.

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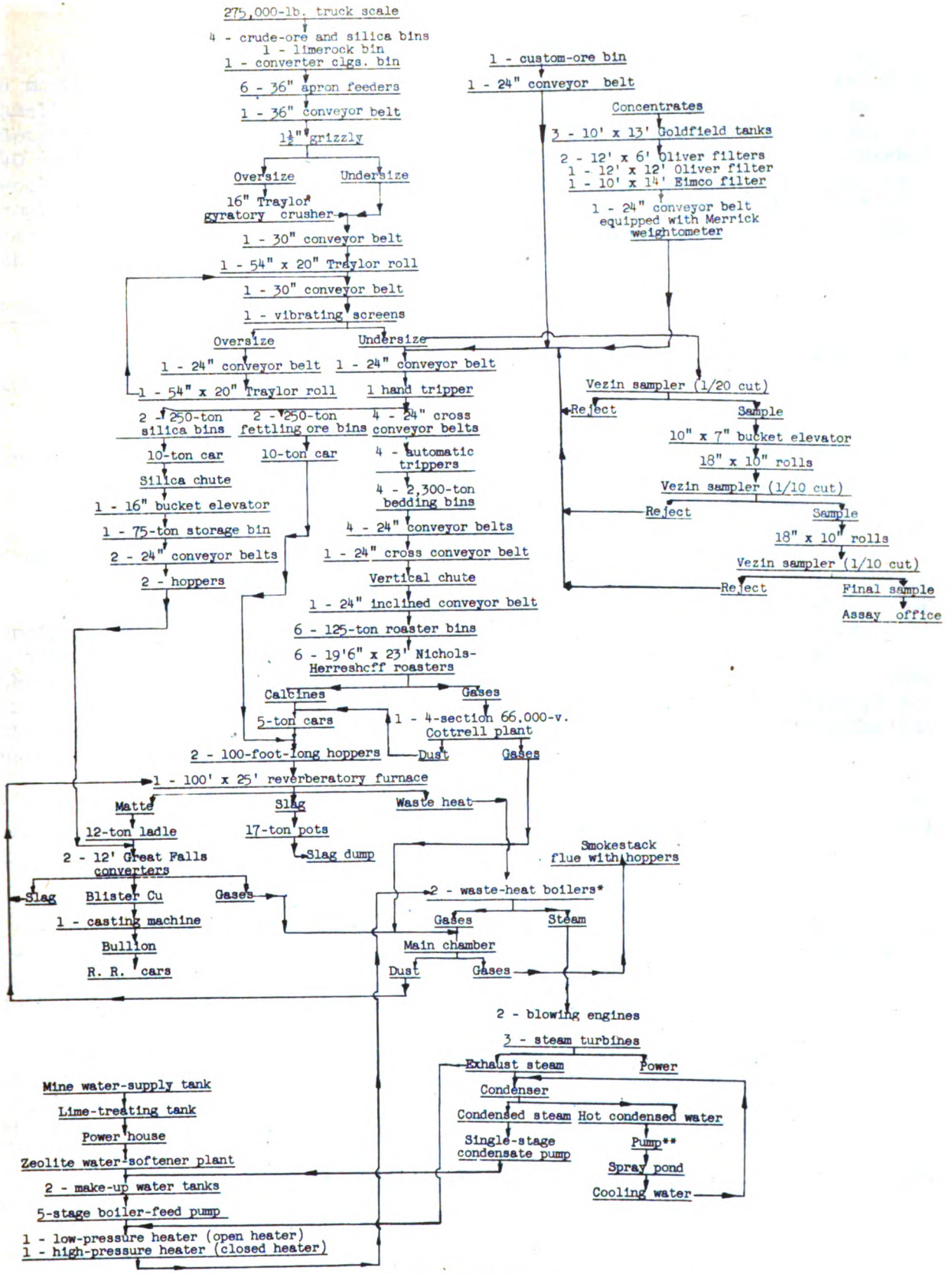

FIGURE 5.- Magma smelter flow sheet.
* 2 direct-fired boilers are used in addition to waste-heat boilers.
**Booster pump available to augment supply of cooling water on No. 3 turbine. This water also used in compressor jackets.

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Loss of copper in the stack is determined by sampling on 4 or 5 days a month by a method devised by the smelter superintendent. The intake of a small blower is inserted in the stack at a point of average velocity as determined by measurement with a Pitot tube. The capacity of the blower is 40 cubic feet a minute. The gas is passed through bags of natural, unwashed wool yarn. After a definite time interval the blower is shut off, and the contents of the bags are weighed. The weight of solids in a cubic foot of gas is determined from the number of feet of gas drawn from the stack, and this multiplied by 250,000 gives the stack loss per minute.

Power

The powerhouse is near the south end of the smelter and approximately 100 feet from the reverberatory furnace.

Two waste-heat boilers and two direct-fired boilers are used. These are supplied by mine water, which is first treated by the zeolite process and then pumped to two make-up water tanks. From these it is pumped by a 5-stage centrifugal-boiler-feed pump through one low-pressure open heater and one high-pressure closed heater, which are supplied with expanded steam from the turbines.

The steam from the boilers goes to three steam turbines and two blowing engines. The exhaust steam passes through a condenser and is then pumped by a single-stage centrifugal pump back to the make-up water tanks. Water for the condenser is drawn from a reservoir by a centrifugal pump. The hot water from the condenser goes to a spray pond for cooling and then is returned to the reservoir. A booster pump is used to increase the volume of cooling water passed through the condenser when necessary.

Air at a pressure of 15 pounds per square inch is supplied by one blowing engine operating at 90 r.p.m. and one blowing machine operating at 4,760 r.p.m. Each machine is rated at 10,000 cubic feet a minute.

Air at a pressure of 100 pounds per square inch is supplied by two compressors operated alternately. One is a 28- by 15- by 21-inch tandem compressor rated at 6,500 cubic feet a minute at 187 r.p.m. and driven by an 800-hp., 2,200-volt, 187-r.p.m. motor. The other is a 34- by 20-1/2- by 24-inch compressor, rated at 4,000 cubic feet a minute at 167 r.p.m.

Two generators driven by steam turbines at 715 r.p.m. supply 1,875 kv.-a. at 2,400 volts. One other generator, driven by a steam turbine at 715 r.p.m., supplies 3,750 kv.-a. at 2,400 volts. These three generators supply all of the power for the smelter and part of that for the mine and mill.

Two 150-kw., 275-volt, 750-r.p.m. motor-generator sets supply the D.C. motors at the smelter only.

Machine Shop

The machine shop handles all heavy work for the mine, mill, and smelter and the locomotives of the Magma Arizona Railroad. The equipment consists of

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one 36-inch lathe, one 24-inch lathe, and one 16-inch lathe; milling machine; shaper; pipe-threading machine; bolt-cutting machine; planer; power saw; universal grinder; metal spray; 300-ton press; sheet-metal rolls; sheet-metal shears; sheet-metal punch; steam hammer; heating furnace; vertical power clamp; radial drill press; arc welding machine; and two acetylene torches. Half of the equipment is driven by individual motors and half by belts from a line shaft.

The shop crew is under the supervision of the master mechanic.

Railroad

The rolling stock of the railroad consists of 1 Consolidation freight locomotive (type 2-8-0), 1 Mogul locomotive (type 2-6-0), ore cars, box cars, and freight cars. The shop crew and railroad are under the supervision of the master mechanic and railroad superintendent.

Assay Office

The assay-office force consists of four chemists and three bucking-room boys. Five balances are used, consisting of three analytical balances, one pulp balance, and one button balance. Self-setting burettes are used for all analytical work. Oil received by the smelter is tested in a centrifuge. The office is equipped with six hot plates. An ultra-violet light is used frequently on the mine samples. Three muffles, burning natural gas at a pressure of 5.0 pounds a square inch in Denver Fire Clay gas burners No. 774, are used for gold and silver assays.

The usual day's work of the assay office includes approximately 75 copper analyses by the iodide method, 30 zinc analyses by Low's method, and 12 iron analyses by the permanganate method, in addition to analyses for sulfur, lime, alumina, silica, and other components of ore.

The bucking-room equipment consists of two jaw crushers, four pulverizers, five drying ovens, one moisture balance, and three Jones splitters. A dust collector is operated in conjunction with ore crushing and pulverizing equipment. Rejects from the bucking room are piled near the railroad track and collected near the close of each year for transfer to the crushing plant. In 1942, 57 tons of this material containing 6.0 percent copper was returned to the smelter.

Oil Tanks

One tank 70 feet in diameter and 30 feet in height, with a capacity of 20,000 barrels, contains oil at all times to provide fuel if the gas supply fails and to furnish oil for the locomotives. Three 200-barrel oil tanks are located near the reverberatory furnace. The oil pump at the large tank is driven by a 10-hp., 440-volt, 715-r.p.m. motor.

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Auxiliary Water Reservoir

A large reservoir on the east side of the smelter and below it is supplied by water from the filter plant and excess water from the mill tailing ponds. Water from this reservoir is used in case the supply of mine water is insufficient. Excess water in this reservoir overflows into Queen Creek.

Labor

The smelter force consists of approximately 150 men, of whom those on operation are all under the supervision of a general foreman.

Filtering

The concentrate from the mill, containing 68 to 75 percent solids, is received at the filter plant in three 13- by 10-foot Goldfield storage tanks placed about 30 feet above the filters. The rakes of these tanks are driven at 5 r.p.m. by a 7-1/2-hp., 440-volt, 715-r.p.m. motor. One sack of lime (60 pounds) is added to each tankful to coagulate the slimes and aid filtering. Material is drawn from one tank at a time and flows by gravity through a 10- by 12-inch launder to the filters. A hole in the bottom of the launder supplies each filter, the quantity being controlled by an adjustable wooden plug.

The filtering equipment consists of two 12- by 6-foot continuous vacuum filters driven by a 7-1/2-hp., 440-volt, 715-r.p.m. motor, one 12- by 12-foot Oliver filter driven by a 5-hp., 440-volt, 715-r.p.m. motor, and one 10- by 14-foot continuous vacuum filter driven by a 3-hp., 440-volt, 715-r.p.m. motor. Agitation at the filters is provided by compressed air at Nos. 1 and 2, by an agitator driven by the filter motor at No. 3, and by an agitator driven by a 2-hp., 440-volt, 1,410-r.p.m. motor at No. 4. Vacuum is induced by three pumps consisting of one 8-valve, 14- by 8-inch pump driven by a 10-hp., 440-volt, 715-r.p.m. motor and two 4-valve, 9-1/2- by 8-inch pumps driven by 5-hp., 440-volt, 720-r.p.m. motors. All vacuum pumps operate at 225 r.p.m.

Water from the filters runs by gravity through a 4-inch pipe to three 12- by 6-foot settling tanks in series. The first tank acts as a barometric seal for the vacuum pumps. The overflow from the last tank runs by gravity to the auxiliary reservoir mentioned above. Solids that have collected in the tanks are cleaned out periodically.

The average rate of filtering is 32 tons an hour, as shown in table 13.

The filter cake, containing 11 to 13 percent moisture, drops onto a 24-inch conveyor belt, parallel to the filters, driven by a 7-1/2-hp., 440-volt, 715-r.p.m. motor. A hand sample is taken from the belt every 10 minutes. An automatic sampler designed by the filter-plant foreman and driven by the belt-idler pullys also samples the material at 61-second intervals. The hand and the automatic samples are split separately and one-half rejected. The remaining half is placed in a can and the final sample taken with an auger.

The load on the belt is weighed by a weightometer located about 20 feet from the head pully.

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This conveyor belt discharges onto the 24-inch inclined cross-conveyor belt, which carries the material from the crushing plant to the bedding bins.

An average analysis of the concentrates for the year 1942 is given in table 8.

Crushing

The crushing plant is at the north end of the smelter, adjacent to the filter plant. The feed to the crushing plant is direct-smelting ore, silica ore, and limerock, which are hauled from the mine by the Magma Arizona Railroad in 50-ton cars. Cleanings from the converter aisle of the smelter are also crushed here. The feed is delivered to seven concrete receiving bins with a capacity of 200 tons each.

The sides and backs of the bins slope toward the feeders, and blowing or picking down is necessary only when the material is wet.

Six bins are used for crude ore, silica ore, converter cleanings, and any custom ore that may be treated, and the seventh is used for custom concentrate or any other material that does not require crushing.

The smelting of custom ore has been of varying importance. Before 1933 only small quantities had been treated, but increase of the price of gold at that time brought in large tonnages of custom ore from operations that had not been profitable at the previous lower price. At present, the production of large quantities of high-silica direct-smelting crude ore from the Magma mine limits the treatment of custom ore to self-fluxing or basic ores, neither of which is often available. Occasionally siliceous copper ores of sufficient grade to warrant direct smelting are purchased for converter flux.

Cars are weighed at the crushing plant on a track scale of 275,000 pounds capacity before and after dumping. Moisture samples, except samples of converter cleanings, are taken from the cars by hand. Material is drawn from each bin by a 36-inch apron feeder operated by a 5-hp., 225-volt, D.C. motor with variable speed of 500 to 1,500 r.p.m. This gives the feeders a speed of 4 to 12 feet a minute. Flow of material is controlled by four swing hammers over each feeder.

The material drops from the apron feeder onto a 36-inch inclined, gravity take-up conveyor belt driven by a 10-hp., 440-volt, 715-r.p.m. motor. Tramp material is removed by belt pickers.

This conveyor delivers the material to a grizzly inclined at 45° with 1-1/2-inch spacing. The oversize passes directly to a 16-inch Traylor gyratory crusher set at approximately 1-1/2 inches and driven by a 75-hp., 440-volt, 720-r.p.m. motor.

The crushed product from the gyratory crusher is joined by the undersize from the grizzly and falls through an inclined chute onto a 30-inch conveyor

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belt driven by a 5-hp., 440-volt, 715-r.p.m. motor. This material is discharged from the belt through an inclined chute into the primary rolls. These are Traylor rolls 54 by 20 inches in size, set at approximately 1 inch and driven at 225 r.p.m. by a 150-hp., 2,200-volt, 290-r.p.m. motor.

The crushed product falls vertically from the rolls onto a 30-inch inclined conveyor belt, in a tunnel beneath the floor, driven by a 10-hp., 440-volt, 715-r.p.m. motor.

The material from the conveyor drops to two vibrating screens with 1/2-inch openings. The screen exciters are operated by one 110-volt, 18.2-amp., 15-cycle, 900-r.p.m. generator rated at 1 kw. and driven by a 2-hp., 440-volt, 710-r.p.m. motor; and one 110-volt, 21.8-amp., 15-cycle, 860-r.p.m. generator rated at 1.2 kw. and driven by a 3-hp., 440-volt, 720-r.p.m. motor.

The oversize from the screens is delivered to a 24-inch inclined conveyor belt driven by a 10-hp., 440-volt, 715-r.p.m. motor, which discharges it through an inclined chute to the secondary rolls, set at approximately 1/2 inch. These rolls are similar to the primary rolls in make, size, and drive.

The product from the secondary rolls falls vertically onto the 30-inch conveyor belt, which carries the crushed product from the primary rolls and thus returns to the screens with which the secondary rolls are in closed circuit.

Undersize from the screens falls to a 24-inch inclined, gravity-take-up conveyor belt driven by a 20-hp., 440-volt, 715-r.p.m. motor, by which it is delivered to the bedding bins.

Dust is drawn from the gyratory crusher and from the tunnel under the primary and secondary rolls by a fan, which is driven by a 7-1/2-hp., 440-volt, 715-r.p.m. motor. The dust is discharged to a cone filled with water, and the material is discharged as a thick mud onto the conveyor which carries the crushed ore to the bedding bins.

A 16-ton hand-operated crane serves the gyratory crusher and rolls.

The average rate of crushing is 62 tons per hour.

Sampling

Feed for the sampling mill is taken from the vibrating-screen feed. A sample of 1.00 pound a ton is taken. The sample mill is driven through line shafts by a 30-hp., 440-volt, 715-r.p.m. motor. The flow sheet of the sampling mill is shown in figure 5.

Bedding

The bedding bins, which are between the crushing and filter plants and the roaster building, are of steel and concrete construction. There are four

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parallel bins, each with a capacity of 2,300 tons. Each bin is 96 feet in length, 27 feet in width, and 18 feet in depth; the sides are inclined at 45° for 12 feet of the height and are vertical for the remainder. The 24-inch conveyor belt mentioned above, which carries the crushed ore from the crushing plant, also carries the filtered concentrate from the filtering plant. A hand tripper discharges material from this belt onto one of four cross-conveyor belts, each over one of the bedding bins. Each cross belt is driven by a 5-hp., 440-volt, 715-r.p.m. motor. When fettling ore or silica ore is conveyed, the hand tripper discharges the material directly into four 250-ton fettling and flux pockets at one end of the bins.

The conveyor belt leading to each bedding bin has an automatic tripper, which runs on tracks above the center of the bin. At each end of the bin is a stop which trips a weighted lever on the carriage and reverses the travel. The tripper is provided with a cross-conveyor belt, which discharges to either side of the center line of the bin and is reversed by the same mechanism that reverses the tripper. Thus the feed is distributed evenly over the surface of the bin. The carriage of the tripper is driven by a 5-hp., 225-volt, D.C., 850-r.p.m. motor, and the cross-conveyor belt is driven by a 3-hp., 225-volt, D.C., 850-r.p.m. motor.

Material is delivered from the bins through a 3-foot slot in the bottom, which is covered with 4- by 8-inch planks. When a bed is ready to be discharged, two or three of these planks are removed at one end of the bin, leaving an opening 16 or 24 inches in width. The material is pulled down by hand by three or four men using large hoes or forks. As the face of the bed retreats the planks which have been removed are replaced, and others are removed next to the face so that the 16- or 24-inch opening is carried along as the work progresses.

The material from the bed drops onto a 24-inch conveyor belt parallel to the bin. This is driven by a 5-hp., 440-volt, 715-r.p.m. motor. It discharges onto a 24-inch cross-conveyor belt, which is parallel to the ends of the bedding bins. This belt also is driven by a 5-hp., 440-volt, 715-r.p.m. motor. Material is discharged from this belt through a chute onto a 24-inch inclined gravity-take-up conveyor, which carries it to the top of the roaster building. The last belt is driven by a 25-hp., 440-volt, 715-r.p.m. motor.

The average content of the beds is 25 to 30 percent crude ore, limerock, and converter cleanings and 70 to 75 percent concentrate. The amount of ore put into a bed is controlled by the composition of the furnace slag and varies inversely with the silica content of the slag. A ratio of 1.5 to 1.6 base : 1 acid is maintained in the beds to give an approximately neutral slag.

Roasting

The feed for the roasting plant is delivered by a hand tripper on the last conveyor mentioned above to six 125-ton roaster bins, each of which feeds a 19-foot 6-inch by 23-foot Nichols-Herreshoff roaster. Each roaster has eight hearths plus a drying hearth. However, only six roasting hearths are used at present, the first two being bypassed.

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The central shafts and rabble arms are cooled by air supplied at a pressure of 3 to 4 ounces a square inch by two fans, each of which is driven by a 2-hp., 440-volt, 715-r.p.m. motor. The hot air is conducted from the top of the central shaft to the bottom hearth, where it is used for combustion. In three roasters the rabbles have bolted flanges, while in the other three they are of a later type and are fastened with pins.

The rabble arms of five roasters are driven by 7-1/2-hp., 440-volt, 715-r.p.m. motors and those of the sixth by a 10-hp., 440-volt, 715-r.p.m. motor.

The hearth arches are conical. The furnaces are constructed of standard fire-clay brick on the inside, except the skew-backs for the arches, in which silica brick is used.

The calcined material is drawn from the roaster hoppers into 5-ton cars with two bottom gates. The gates of the hoppers are operated by levers controlled by the motor crew from outside the roaster building.

The calcine cars are weighed by the motor crew, before and after loading, on a track scale of 60,000 pounds capacity and are then hauled on standard-gage track by battery locomotives to the reverberatory furnace bins. At the furnace the track divides, and a branch runs along each side of the furnace about 12 feet above the arch. The calcines are dropped into hoppers under the tracks.

58 to 60 percent of the sulfur in the feed is removed by roasting. The average analyses of charge and calcines in 1942 are given in table 11.

The roaster temperature is about 1,200° F. and the consumption of gas 115 cubic feet per ton of material roasted.

The average rate of roasting in 1942 was 83 tons per roaster per day. However, the roasters can be crowded to treat more than 100 tons per day.

Cottrell Plant

The roaster gases are led to a Cottrell plant for the collection of dust and are the only feed for this plant. The plant is of the pipe type, has four sections, and is operated at 66,000 volts. The power plant contains five motor generators. Two D.C., 4-kw. exciters are used, one at a time, to excite two of three 50-cycle A.C. generators. The exciters are driven by 7-1/2-hp., 440-volt, 1,435-r.p.m. motors. Each A.C. generator is driven by a 40-hp., 440-volt, 1,435-r.p.m. motor.

Approximately 85 percent of the dust is removed. The gas from the Cottrell plant passes through the main chamber, which is provided with hoppers, and then to the stack.

An analysis of the dust is given in table 9.

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Smelting

The reverberatory furnace is 100 feet in length and 22 feet in inside width, with over-all height of 13 feet 11 inches. The bottom 4 feet is filled with crushed silica. The side walls are built of silica brick 27 inches thick for the first 5 feet 10 inches, 22-1/2 inches thick for the next 5 feet, and 18 inches thick for the remainder of the height. The arch is built of silica brick, 20 inches thick in the panels and 27 inches in the ribs; the radius is 29 feet 3 inches. The uptake arch has a radius of 9 feet and an inside width of 8 feet.

Temporary repair, by brickwork or guniting, is done from time to time. Gunite is used only on thin spots to make them last as long as thicker adjacent sections. The furnace is shut down every 6 or 7 months, or whenever the entire arch has become too thin.

The furnace is fired by four natural-gas burners of Venturi type. The gas is supplied at a pressure of 18 to 20 pounds a square inch. This pressure is reduced to 5 pounds a square inch during charging of the furnace to prevent the blowing of fine material against the arch, with consequent fluxing of the brick. The furnace temperature ranges from 2,400° to 2,600° F. A furnace draft of 0.09 inch of water is maintained by manually controlled dampers on the waste-heat boilers. The average consumption of gas is 4,091 cubic feet a ton of solid charge.

Each side of the furnace has 24 charging or fettling holes spaced at intervals of 3 feet 6 inches. The last hole on each side, near the uptake, is not used. Calcines are charged to the furnace through the 12 holes nearest the firing end on each side. Siliceous fettling material, which is mainly siliceous ore from development headings in the mine, is charged through 11 of the 24 holes on each side, and the sides are kept well-fettled. The depth of the bath is maintained at 24 to 27 inches.

The matte tap hole is 7 inches above the crushed silica bottom and about 30 feet from the uptake end. When new, tap holes are 1-1/4 inches in diameter, and they are used until the diameter becomes too great, the largest diameter used commonly being about 3-1/2 inches. A tap hole will last 20 to 30 days. An oxygen blowpipe is used whenever difficulty is experienced in opening a new tap hole or whenever a tap hole freezes.

The skimming hole for the slag is 18 inches above the silica bottom or 11 inches above the level of the tap hole.

At present, the charge per shift is 32 cars (approximately 160 tons) of calcine, 1 to 2 cars (5 to 10 tons) of Cottrell dust, 2 to 3 cars (10 to 15 tons) of siliceous fettling material, and 1 to 2 cars (5 to 10 tons) of main-chamber dust. The total charge per shift is approximately 190 to 200 tons, from which are produced 6 to 8 ladles (72 to 96 tons) of matte and 8 to 10 pots (136 to 170 tons) of slag.

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The matte, which has a grade of 32 to 34 percent copper, runs through a semicircular cast-iron launder into matte ladles of 12 tons capacity. A 40-ton crane carries the ladles to the converters.

The slag runs into a launder built of crushed silica, then into a semicircular cast-iron launder running through the floor, and into the motor-dump cinder cars on tracks below.

The capacity of these cars or pots is 225 cubic feet - approximately 17 tons. They are hauled by a trolley locomotive to the slag dump south of the smelter.

Matte and slag are sampled by hand during each tapping or skimming. Samples are taken with long spoons and granulated by pouring into water. The slag dump is sampled by hand once a day to check furnace samples. Average analyses of charge, matte, and slag for 1942 are given in table 11.

The furnace is operated by three men under the supervision of a furnace foreman.

Converting

The converting plant consists of two 12-foot Great Falls converters lined with 15-inch magnesite brick all around the cap and on the tuyère side of the shell. They were designed for a brick lining on the bottom, but a layer of slag has been proved to be satisfactory and is now used. The mouth of the converter is 6 feet 6 inches in diameter. The gases escape into a hood and pass through a flue equipped with hoppers. Air is supplied at the rate of 10,000 cubic feet a minute and at a pressure of 15 pounds per square inch by two blowing engines in the power house. There are 24 tuyères with 1-1/2 inches inside diameter.

Siliceous converter flux is hauled from the silica pocket at the bedding bins in a 10-ton car and dumped into a chute beneath the tracks and above the converters. From this chute the flux is fed to a bucket elevator driven by a 7-1/2-hp., 440-volt, 715-r.p.m. motor and lifted to the top of a 75-ton storage bin. A conveyor-belt feeder on each side of the storage bin delivers the flux into a hopper just above each converter, from which a pipe leads to the mouth of the converter. The feeders are driven by 3-hp., 440-volt, 715-r.p.m. motors, and the motors and the gates on the hoppers are controlled by the converter operators.

Each converter is started by the charging of one ladle (12 tons) of matte. Silica is then added, and the converter is turned up and blown for about 45 minutes. The slag formed is then poured off, one more ladle of matte and additional flux are added, and the converter is again blown until the matte is near the white-metal stage, as shown by the rabble and the converter gases. The converter foreman signals to the reverberatory foreman by a compressed-air whistle when a charge of matte is wanted. The slag is then poured off, and the mattes from the two converters are combined in one converter, with any

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scrap copper that has accumulated, and blown to finished copper. The actual blowing time is about 20 minutes a ton of copper produced. The converter slag is poured back into the head end of the reverberatory furnace. Converter slag is sampled by hand from the ladles after the slag has been poured off. Table 14 gives an analysis of the slag.

The average rate of converting in 1942 was 11.9 tons of bullion per charge. An analysis of the bullion is given in table 10.

When a hot spot occurs in one of the converters, the contents are transferred to the other converter, and a mixture of crushed magnesite brickbats, finely ground magnesite, and water glass is thrown into the converter to coat the hot spot. This is allowed to set for approximately 30 minutes before the converter is again put into service.

Any converter-aisle cleanings which are not charged to the converter are picked up in "boats" and dumped by the crane onto an elevated horizontal grizzly beneath which a railroad car is spotted. A heavy iron "ball" breaks any large pieces left on the grizzly. Water sprays are used to cool any heated scrap and minimize the dust. When the railroad car is filled it is hauled to the bins at the crushing plant.

The converter gases join the reverberatory gases and pass through a flue to the smelter smokestack. This flue is equipped with hoppers, and any solid material that may accumulate is cleaned out during the shut-downs. The dust chamber just ahead of the smokestack is cleaned out at the same time.

The converter aisle is served by one of two 40-ton cranes. The main hoist is driven by a 75-hp., 225-volt, D.C., 450-r.p.m. motor; the bridge by a 60-hp., 225-volt, D.C., 450-r.p.m. motor; the carriage by a 15-hp., 225-volt, D. C., 450-r.p.m. motor; and the auxiliary hoist by a 22-1/2-hp., 225-volt, D.C., 450-r.p.m. motor.

Casting

The finished copper is transferred by a 12-ton ladle from the converter to the casting machine. The casting machine is of the link-belt or straight-line type, consisting of a series of copper molds carried by an endless link belt. The molds are cast from low copper from the converters. Milk of lime is put into the bottom to prevent sticking of copper bars. Cracked molds are thrown back into the converters.

The ladle which contains the blister copper is set on a tilting mechanism at one end of the casting machine and elevated along the arc of a circle to provide a constant flow. This pouring mechanism is driven by a 30-hp., 440-volt, 675-r.p.m. motor. The copper flows into a spoon-shaped receptacle tilted by a hydraulic lift operated by a 3-hp., 440-volt, 715-r.p.m. motor. Continuous flow of material prevents crusting over in the spoon. The link belt, which is driven by a 25-hp., 440-volt, 690-r.p.m. motor, passes directly underneath the spoon.

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All of the mechanisms are controlled from a protected booth at the side of the pouring end by the casting-machine operator. Water sprays cool the copper before it reaches the end of the machine. The copper bars are pried loose from the molds as they pass over the pully at the end and are taken by hand trucks to platform scales where 10 bars are weighted at a time and are then loaded into railroad cars at the side of the scales.

The bullion is sampled with a doughnut-shaped sampler by a member of the bullion gang. The samples are taken to the machine shop, where they are sawed into quarters with a hacksaw; the cuttings are sent to the assay office.

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TABLE 1. - Analyses of Concentrator products for 1942

Product	Weight, percent	Copper, percent			Zinc, percent			Percent					Ounces a ton	
		Total	Sul.	Ox.	Total	Sul.	Ox.	Fe	CaO	Al_2O_3	SiO_2	S	Au	Ag
Tables:														
Heads	100.00	5.122	5.102	0.020				12.7	1.5	10.6	50.0	15.1	0.023	1.043
Conc.	12.80	11.30	-	-				33.8	.9	2.5	7.6	41.9	.060	2.00
Tailing	87.20	4.25	-	-				10.1	1.5	10.8	55.4	11.7	.010	1.00
Flotation:														
Heads	93.60	4.25	-	-				10.1	1.5	10.8	55.4	11.7	.010	1.0
Conc.	26.66	13.21	-	-				27.2	.9	3.2	12.6	36.8	.068	2.64
Tailing	66.94	.275	-	-				4.4	1.7	11.6	67.8	4.2	.003	.115
Zn flotation:														
Heads	100.00	1.181	-	-	8.412	8.072	0.340	12.5	1.3	3.5	42.0	13.5	.015	2.325
CRC 1/	12.51	8.139	-	-	12.204	-	-	27.1	.8	.9	3.6	40.1	.060	12.0
CRT	87.49	.310	-	-	7.944	-	-	11.0	1.5	3.7	46.0	10.3	.007	1.232
ZCC	12.10	1.339	-	-	49.575	-	-	6.6	.9	1.0	4.6	28.6	.030	5.753
ZRT	75.39	.198	-	-	1.776	-	-	11.2	1.7	3.9	52.4	7.3	.005	.592
Over-all milling:														
Heads	100.00	4.239	-	-	1.606								.021	1.359
Cu Conc.	31.73	12.615											.067	3.362
Tailing	68.27	.260											.004	.263

1/ CRC - Copper rougher conc.
CRT - Copper rougher tail.
ZCC - Zinc cleaner conc.
ZRT - Zinc rougher tail.

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TABLE 2. - Moisture of mill heads for 1942

Ore	Moisture, percent
Copper	2.990
Zinc	3.174

TABLE 3. - Mill recoveries for 1942

Product	Recovery, percent				
	Cu	Fe	Au	Ag	Zn
Tables	28.45	26.75	40.75	24.45	
Flotation	95.83	72.55	108.77	94.09	
CRC[1]	76.67	23.72	57.86	49.34	16.66
ZCC[2]	14.14	6.37	24.24	29.94	70.35
Over-all	95.42	38.60	103.94	88.54	70.45

1/ Copper rougher conc.
2/ Zinc cleaner conc.

TABLE 4. - Concentration data for 1942

Copper ore milled	tons	281,283
Zinc ore milled	do.	87,105
Copper ore treated by tables	do.	192,278
Table concentrate produced	do.	21,780
Copper ore treated by flotation	do.	259,503
Copper flotation concentrate produced	do.	104,545
Copper flotation tails	do.	176,738
Recovery of copper in copper ore	percent	95.827
Copper concentrate produced from zinc ore	tons	12,258
Tails produced from selective flotation of copper	do.	74,847
Recovery of copper from zinc ore	percent	76.735
Zinc lost in copper concentrate	do.	23.113
Zinc concentrate produced	tons	8,741
Zinc tailing produced	do.	66,106
Recovery of zinc in zinc circuit only	percent	86.205
Recovery of zinc over-all	do.	70.45
Zinc lost in zinc tails	do.	13.438
Ratio of concentration:		
Tables		7.825
Copper flotation		3.510
Selective copper flotation		7.106
Over-all zinc flotation		9.965
Zinc circuit only		8.563
Final copper		2.691
Ball consumption:		
5-inch	lb. per ton Cu ore	1.149
Do.	lb. per ton Zn ore	.447
4-inch	do.	.683
3-inch	lb. per ton Cu ore	.031
Do.	lb. per ton Zn ore	.216

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TABLE 4. - Concentration data for 1942 (Cont'd)

2-inch	lb. per ton Cu ore	1.705
Do.	lb. per ton Zn ore	1.991
Total	lb. per ton total ore	2.992
Lime consumption	lb. per ton Cu ore	3.910
Do.	lb. per ton Zn ore	6.180
Total reagent consumed	lb. per ton Cu ore	.403
Do.	lb. per ton Zn ore	4.595
Copper flotation-feed pulp	percent solids	21.06
Selective copper flotation-feed pulp	do.	20.95
Zinc flotation-feed pulp	do.	15.71
Copper flotation-feed alkalinity	lb. CaO per ton H_2O	.333
Zinc flotation-feed alkalinity	do.	.224
Dorr-thickener underflow	percent solids Cu conc.	66.12
Do.	percent solids Zn conc.	66.18
Copper flotation tailing	percent through 65-mesh	92.90
Zinc flotation tailing	do.	98.50
Copper flotation air pressure	lb. per sq. in.	1.500
Zinc flotation air pressure	do.	1.250
Flotation pH:		
Copper flotation		10.81
Selective copper flotation		6.30
Zinc flotation		10.80
Power consumption:		
Copper ore	kw.-hr. per ton	22.67
Zinc ore	do.	20.64

TABLE 5. - Summary of milling costs for 1942

	Total	Cost per ton
Callow screens	$1,286.70	$0.004
Conveyors	1,245.59	.003
Drag classifiers	2,170.62	.006
Elevators	3,364.67	.009
Launders	625.93	.002
Primary grinding	54,695.69	.148
Secondary grinding	39,639.77	.108
Blowers	3,884.86	.011
Lime	16,821.64	.046
Reagents	47,661.41	.129
Pumps	8,814.75	.024
Agitair cells	16,789.09	.045
Lights	1,006.79	.003
Motors	3,083.14	.008
Telephones	19.76	.000
Substation	15.27	.000
Power transmission line	36.94	.000
Compressed-air lines	35.75	.000
Compressed air	2,390.67	.007

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TABLE 5. - Summary of milling costs for 1942 (Cont'd.)

	Total	Cost per ton
Fire protection	$105.83	$0.000
Mill cleaning	7,877.83	.021
Misc. labor and supplies	4,051.73	.011
Oil and waste	7,968.93	.022
Pullys, shafts, and belts	705.84	.002
Repairs mill buildings	344.31	.001
Shop	1,501.09	.004
Samplers	3,365.25	.009
Akins classifiers	2,124.23	.006
Dorr thickeners	1,338.18	.004
Settling tanks (Goldfield)	1,192.92	.003
Concentrate filtering and sampling	3,062.77	.008
Superintendence	28,767.28	.078
Tailings dam	12,231.23	.033
Tables	6,546.60	.018
Water supply	4,032.72	.011
Watchmen	6,727.91	.018
Coarse crushing	17,723.02	.048
Mill stores shortage	246.89	.001
Proportion of gen. overhead	39,195.36	.106
Total milling costs	352,696.96	0.957

TABLE 6. - Direct cost of operating section 1 zinc mill, 1942

	Total	Cost per ton
Compressed-air lines	$9.33	$0.000
Mill cleaning	282.15	.003
Misc. labor and supplies	1,144.77	.013
Samplers	42.46	.000
Water supply	718.69	.008
Motors	478.39	.006
Conveyors	110.71	.001
Drag classifiers	143.66	.002
Elevators	195.10	.002
Launders	263.56	.003
Primary mill	11,495.88	.132
Secondary mill	5,769.79	.066
Blowers	1,716.96	.020
Lime	4,776.41	.055
Reagents	27,324.12	.314
Pumps	2,201.93	.025
Agitairs	3,965.74	.046

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TABLE 6. - Direct cost of operating section 1 zinc mill, 1942 (Cont'd.)

	Total	Cost per ton
Akins classifiers	$145.16	$0.002
Dorr thickener	823.13	.009
Settling tank (Goldfield)	165.01	.002
Filtering and sampling	3,062.77	.035
Tailings dam	258.10	.003
Coarse crushing	2,085.01	.024
Total direct cost	67,173.83	.771

TABLE 7. - New material treated and new material received at smelter for 1942

		Analysis		
New material received	Tons	Copper, percent	Silver, oz.	Gold, oz.
Crude ore	80,758	8.238	2.024	0.043
Concentrate	117,126	12.598	3.363	.070
Limerock	9,147			
Total new mat'l. rec'd.	207,031			
Converter byproducts	17,484	16.044	4.177	.077
Flue dust	395	32.292	9.621	.110
Brickbats	182	1.662	.377	.003
Total received	225,092			
New material treated				
Crude ore	82,159	8.264	2.038	.043
Concentrate	118,073	12.604	3.379	.070
Limerock	9,217			
Total new mat'l. treated	209,449			
Converter byproducts	17,899	16.013	4.148	.076
Flue dust	395	39.292	9.621	.110
Brickbats	182	1.662	.377	.003
Total treated	227,916			

TABLE 8. - Roaster statement for 1942

		Analysis		
	Tons	Copper, percent	Silver, oz.	Gold, oz.
Crude ore	41,397	9.508	2.281	0.049
Concentrate	118,073	12.604	3.379	.070
Limerock	9,164			
New mat'l. treated by roasters	168,634			

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TABLE 8. - Roaster statement for 1942 (Cont'd.)

	Tons	Analysis: Copper, percent	Analysis: Silver, oz.	Analysis: Gold, oz.
Converter byproducts	3,754	13.634	3.422	0.063
Flue dust	395	39.292	9.621	.110
Total treated by roasters	172,783			
Roaster products				
Calcine and gravity dust	149,962	12.519	3.241	.062
Cottrell dust	6,107	14.097	4.981	.049
Roaster flue dust	1,436	10.200	6.100	.080
Total products	157,505			

TABLE 9. - Reverberatory and converter statement for 1942

	Tons	Analysis: Copper, percent	Analysis: Silver, oz.	Analysis: Gold, oz.
Calcine and gravity dust	149,962	12.519	3.241	0.062
Cottrell dust	6,107	14.097	4.981	.049
Main-chamber dust	1,209	10.200	6.100	.080
Converter flue dust	724	59.461	12.951	.209
Fettling ore	19,638	7.008	1.802	.036
Converter cleanings	14,136	16.645	4.341	.080
Limerock	53			
Total solid charge	191,829			
Converter flux	21,124	6.994	1.780	.036
Brickbats	182	1.662	.377	.008
Total charge	213,135			
Products				
Bullion	20,530	98.982	27.788	.649
Reverberatory slag	153,767	.577	.103	.003
Reverberatory flue dust	332	12.600	6.659	.060
Converter flue dust	850	59.555	12.926	.209
Converter byproducts	17,484	16.044	4.177	.077
Total products	192,963			

TABLE 10. - Smelter products and recoveries for 1942

	Tons	Analysis: Copper, percent	Analysis: Silver, oz.	Analysis: Gold, oz.
Bullion	20,530	98.982	27.788	0.649
Reverberatory slag	153,767	.577	.103	.003
Stack loss	6,439	8.083	3.278	.040
	180,736			

Metal recoveries	
Cu	93.816
Ag	100.663
Au	113.840

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TABLE 11. - Average analyses of smelter products for 1942

	Analysis, percent					
	Cu	Fe	SiO_2	Al_2O_3	CaO	S
Roaster charge	11.28	24.7	18.1	4.2	3.4	29.7
Calcines	12.59	26.9	23.7	5.1	4.7	10.6
Reverberatory slag	.51	34.2	38.0	8.2	4.5	
Converter slag	2.14		22.1			
Matte	31.69	34.2				22.4

TABLE 12. - Fuel statement for 1942

	Natural gas, millions cubic feet	Millions cubic feet per ton
Roasters	19,814	0.115
Reverberatory	784,733	4.091
Boilers	264,751	---
Total	1,069,298	

TABLE 13. - Smelter statistics for 1942

Average rate of crushing	62	tons per hour
Average rate of filtering	32	tons per hour
Average rate of reclaiming	68	tons per hour
Average rate of roasting	83	tons per hour
Average rate of smelting	556	tons solid charge per day
Average rate of converting	11.9	tons bullion per charge
Average steam production	31,524	lb. per hour (waste-heat boilers)
Average equiv. evaporation	336	lb. per millions cubic feet burned (waste-heat boilers)
Average equiv. evaporation	664	lb. per millions cubic feet burned (direct-fired boilers)
Average turbine load	2,358	kw.

TABLE 14. - Complete analyses of smelter feed and products for 1942

	Analysis, percent					
	Cu	SiO_2	Fe	CaO	Al_2O_3	S
Filtered concentrates	13.72	11.17	29.66	0.63	3.35	37.42
Crude ore plus silica ore	8.41	44.80	14.52	1.07	8.30	16.92
Limerock	.37	7.17	1.10	50.15	1.05	
Calcines	12.48	24.20	27.20	4.60	5.06	10.66
Matte	31.69		34.20			22.40
Reverberatory slag	.50	38.40	33.76	4.58	8.28	
Converter slag	2.08	22.40	49.70	.68	3.76	

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